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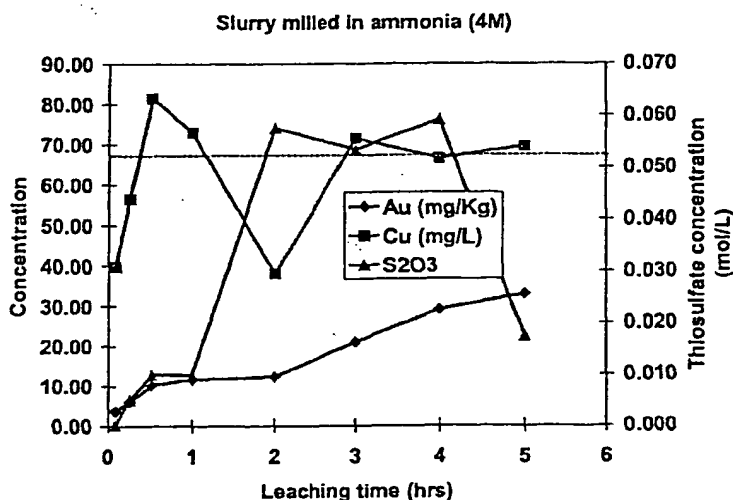
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(54) Title: PROCESS FOR LEACHING PRECIOUS METALS



(57) Abstract: A process for leaching precious metals such as gold from an ore such as a sulphide ore, the process including the steps of milling an ore containing a precious metal to produce a milled ore, contacting the milled ore with water, an alkaline reagent, a suitable source of oxygen and a suitable source of sulfur (which may be the ore itself) in amounts sufficient to generate at least 0.02 mol/L thiosulfate in situ, and leaching the milled ore into a thiosulfate containing aqueous phase. The thiosulfate generated in situ may constitute at least 30% of the thiosulfate required to effect precious metal extraction into the aqueous phase. The process enables a significant proportion or all of the thiosulfate required to effect leaching of the precious metal to be generated in situ, thus reducing or avoiding the cost and potential difficulties associated with the separate addition of thiosulfate solution.

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PROCESS FOR LEACHING PRECIOUS METALSBackground

The present invention relates to a process for
5 leaching precious metals from an ore.

Gold is an example of a precious metal which may
be extracted from an ore by dissolution into an aqueous
phase containing a reagent that reacts with the gold.
Cyanide is one of the best known reagents or lixivants for
10 extracting or leaching gold into an aqueous phase.

Interest in the use of non-cyano methods for the
dissolution of gold arises from concerns regarding the
toxicity of cyanide. This concern has been heightened
recently by well-publicised accidents in Romania and Papua
15 New Guinea.

In addition to cyanide being hazardous, metal
cyanide species derived from cyanide leaching, eg. copper
cyanide and other cyanide species end up in the tailings
dam. This can lead to environmental hazards of the type
20 experienced in Romania, when cyanide pollutants reach
either the water table or escape into the river system.

Thiosulfate has been proposed as an alternative
to cyanide as a lixiviant for gold from auriferous ores.
Processes that incorporate the use of thiosulfate involve
25 the direct addition of a thiosulfate leach liquor, and
usually pressure leaching of the valuable metal from the
ore. Other processes that incorporate the use of
thiosulphate require high levels of alkali and/or oxygen to
chemically react with the ore to release valuable metals
30 entrapped in the ore to make the valuable metal available
for leaching. The high levels of alkali required add
significantly to the cost of such processes.

It is an object of the present invention to
provide an alternative process in which one or more of
35 these problems are avoided.

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Summary of the Invention

The present invention is based on the discovery that sufficient thiosulfate ions to leach precious metals can be generated in situ when a particulate ore such as a sulfide ore is exposed to a suitable source of oxygen in an alkaline aqueous environment.

The present invention therefore enables a significant proportion or all of the thiosulfate required to effect leaching of the precious metal to be generated in situ, thus reducing or avoiding the cost and potential difficulties associated with the separate addition of thiosulfate solution. Following from this, a number of embodiments of the invention are envisaged.

According to a first embodiment, the present invention provides a process for leaching precious metals from an ore, the process including the steps of milling an ore containing a precious metal to produce a milled ore, contacting the milled ore with water, an alkaline reagent, a suitable source of oxygen and a suitable source of sulfur in amounts sufficient to generate at least 0.02 mol/L thiosulfate in situ, and leaching the milled ore into an aqueous thiosulfate phase.

The amount of 0.02 mol/L thiosulfate being generated in situ can be supplemented by the separate addition of thiosulfate solution, although it will be appreciated that significant advantages flow from the absence of this separate addition. Although in a typical example of the process 0.05 mol/L of thiosulfate will be generated in situ, in fact, it has been found that up to 0.14 mol/L of thiosulfate can be generated in situ. This is significantly in excess of the thiosulfate concentration required to effect precious metal extraction into the aqueous phase. Preferably, the thiosulfate generated in situ constitutes from 30% to 100% of the thiosulfate present in the aqueous phase, more preferably 50% to 100% and most preferably 80% to 100%.

According to a second embodiment, the present

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invention provides a process for leaching precious metals from an ore, the process including the steps of milling an ore containing a precious metal to produce a milled ore, contacting the milled ore with water, an alkaline reagent, a suitable source of oxygen and a suitable source of sulfur in amounts sufficient to generate thiosulfate in situ to form a thiosulfate-containing aqueous phase, and leaching the milled ore into said aqueous phase, wherein the thiosulfate generated in situ constitutes at least 30% of the thiosulphate present in the aqueous phase.

Following leaching of the precious metal, the precious metal is usually recovered from the aqueous phase by any suitable means. Consequently, the process is typically one for the leaching and recovery of the precious metal, and involves the further step of recovering the precious metal from the aqueous phase.

Brief Description of the Drawings

Figure 1 is a graph illustrating the concentration of gold extracted per kilogram of ore during milling of a pyrite concentrate in the presence of ammonia solution.

Figure 2a is a graph of precious metal concentration versus leaching time.

Figure 2b is a graph of thiosulfate and polythionate concentration in solution versus leaching time.

Figure 3a is a graph of thiosulfate concentration and gold leaching versus leach/milling time in the presence of oxygen and 0.3M NaOH.

Figure 3b is a graph of thiosulfate concentration and gold leaching versus leach/milling time in the presence of 0.4M ammonia-oxygen and 0.15M NaOH in solution.

Figure 4 is a graph of precious metal concentrations in the leach liquor, for each of the three leach stages.

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Figure 5 is a graph of thiosulfate generated as a function of available alkalinity under different atmospheric conditions (oxygen, air or nitrogen atmosphere).

5

Detailed Description of the Invention

Various aspects of the invention will now be described in further detail.

10 In contrast to the prior art, the alkalinity of the aqueous phase, due to the presence or addition of alkali, can be kept relatively low. This is an advantage as costly addition of high amounts of alkali can be avoided. Typically, the pH of the aqueous phase is between 7 and 11. At a pH above 11, any copper that may be present
15 may start to precipitate out of solution. Of course, if copper is not present, then the pH may be a little higher, although this is not preferred. In a typical embodiment of the invention, the pH will be in the range of 9.5 to 10.5. At this pH range, oxidation or excessive oxidation of
20 thiosulfate to trithionate and tetrathionate is reduced.

Any suitable alkaline reagent may be used, such as ammonia, lime, sodium hydroxide, potassium hydroxide, soda ash or potash, or a mixture thereof.

It is also advantageous to include an agent that
25 reduces the passivation of gold and stabilises cuprous ions in solution. Such agents control the activity (i.e. solution eH) of copper in the ore, and thus reduce reagent consumption. Ammonia and amino-acids are examples of such reagents. If ammonia is used as the alkaline reagent, it
30 provides this dual function. An example of an amino acid reagent is glycine, although it is noted that any compound containing amine and carboxylic acid functional groups are within the scope of this term.

The source of oxygen may be elemental oxygen in
35 the form of oxygen itself or a gas mixture containing elemental oxygen such as air. Alternatively, the source of oxygen may be sulfur dioxide, hydrogen peroxide or another

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suitable oxidant. Typically, the oxygen source is required to provide a dissolved oxygen content in the aqueous phase of 1 mg/L, and up to 20 mg/L. Usually, the oxygen content will be at the lower end of this range, at around 2 - 4
5 mg/L. Oxygen will be taken up in side reactions occurring in the leaching process. Therefore sufficient oxygen is required to meet those needs, and to produce the thiosulfate in situ, without being so excessive as to cause excess oxidation of the thiosulfate to tetrathionate.

10 For the avoidance of any doubt, it is noted that the term "ore" encompasses ore concentrates. Although the ore is suitably a sulfide ore, other precious metal-containing ores such as oxide ores may be subjected to the process of the invention.

15 The source of sulfur may be a sulfide ore that may or may not contain precious metals. In the case where the ore being treated is not a sulfide ore, the source of sulfur may be elemental sulfur. Preferably however the ore containing the precious metal is a sulfide ore. Recovery of
20 precious metals from a sulfide ore may be enhanced by adding to the fine particles of milled sulfide ore a suitable source of sulfur such as elemental sulfur.

The term "precious metal" encompasses all precious metals that may be leached with thiosulfate, and
25 includes, at a minimum, gold, silver, platinum and palladium. These metals may be present in the ore as elemental gold or in solid solution. This applies particularly to gold.

Copper acts as a catalyst by increasing gold
30 dissolution rates in the presence of thiosulfate. Without wishing to be confined to any particular theory, it appears that cuprous ions oxidise elemental gold to form a gold thiosulfate complex in the presence of thiosulfate ions. Consequently, in one embodiment of the invention, the
35 leaching is conducted in the presence of copper, preferably cuprous ions.

Although the leaching step of the present

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invention may take place at a temperature of up to 70°C, it can be performed at slightly elevated temperatures (up to 50°C) or even at ambient temperature. This is a significant advantage, that can lead to cost savings. In addition, although the leaching step can be conducted at slightly elevated pressures, it is suitably conducted at atmospheric pressure. Conducting the leaching process at ambient temperature and at atmospheric pressure reduces the complexity of the plant design.

10 Milling of the ore can take place contemporaneously with the leaching in any suitable equipment such as an attrition mill (i.e. stirred mill) or a ball mill.

15 Milling is a relatively expensive operation, and therefore it is envisaged that some or all of the leaching stage will not take place contemporaneously with the milling. Milling to the appropriate particle size can be conducted relatively quickly, for instance within about 2 minutes. Thereafter the leaching step takes from 1 hour and up to 48 hours, preferably from 1 to 24 hours, depending on the conditions.

20 The milling of the ore is preferably conducted to produce a relatively fine ore. The ore is suitably ground to a particle size having a P_{80} of 100 micrometres or less. P_{80} refers to 80% of the particles being under 100 micrometres in size. The finer the ore is milled the greater the release of precious metals for leaching. Consequently, a finer particle size is preferred, namely P_{80} 75 micrometres, more preferably P_{80} less than 50 micrometres, and most preferably, less than 20 micrometres. The degree of grinding will depend on the ore itself, and other components of the process. In the case of sulfide ores, the finer the ore is milled, the more reactive is the sulfide to generating thiosulfate in addition to the greater the release of precious metals for leaching. With the development of attrition mills and the like, the reduced cost of fine milling of ores can make the milling

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process an efficient alternative pre-treatment process to those such as roasting or pressure leaching.

The recovery of the precious metal from the ore may be achieved by any of the techniques known in the art, such as ion exchange, precipitation and so forth.

Whilst the process of the invention may be carried out in batches or in a continuous process, in most commercial operations the process will be conducted continuously. Persons skilled in the art will be able to establish an appropriate arrangement including suitable types of vessels, recirculation streams, and multiple stages for conducting each of the process steps, taking into account the basic principles taught in the present application.

In the claims which follow and in the preceding description of the invention, except where the context requires otherwise due to express language or necessary implication, the word "comprise" or variations such as "comprises" or "comprising" is used in an inclusive sense, i.e. to specify the presence of the stated features but not to preclude the presence or addition of further features in various embodiments of the invention.

Various embodiments of the invention are described in further detail below with reference to the Figures and Examples. The examples all utilised the same ore. Mineralogical analysis showed the gold present in the ore existed as fine-grained inclusions in pyrite, as both coarse and fine-grained tellurides locked in and around pyrite grains, and as coarse and fine free milling gold. To enable comparison to cyanide-based extraction processes, it was established that approximately 65% of the gold could be extracted using the conventional cyanidation method, with high reagent consumption.

Example 1

This example demonstrates milling being conducted contemporaneously with leaching/extraction.

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A gold copper sulfide concentrate comprising mainly pyrite (52%), quartz (15%), muscovite (11%), chlorite (2%), albite (5%), ankerite (12%), and traces of rutile (1.4%) and paragonite (0.5%) was ground in a batch
5 1.4 litre attritor mill for 5 hours under mixtures of either ammonia-oxygen or ammonia-SO₂ in solution. The attritor mill comprised a grinding chamber in which the vertical shaft of an impeller with a series of outwardly directed stirring arms, was rotated by means of a drill
10 press. Millings were carried out using 1-1.2 mm ceramic media, a solids to water ratio of 1:2, a ball to sample weight ratio of 5:1, and an impeller tip speed of 9.3 m/s. In all experiments, the pH was kept constant at a value of 10 using ammonia solution. For the ammonia-oxygen milled
15 leach, oxygen flow into the grinding chamber of the mill was maintained at 0.5L/min during milling. This corresponded to a dissolved oxygen level of between 2 and 4ppm during the principal stage of extraction. The ammonia-sulfur dioxide experiments were carried out by bubbling SO₂
20 gas into ammonia solution prior to the addition of ore and subsequent milling.

Figure 1 illustrates the concentration of gold extracted per kilogram of ore during milling of a pyrite concentrate in the presence of ammonia solution and oxygen.
25 Copper which catalyses gold dissolution is dissolved from the ore. Thiosulfate concentration reached a maximum of 0.06mol/L.

The horizontal line in Figure 1 represents the head grade of ore.

30 The calculated head, residue and % gold extraction for three experiments after five hours of milling under different milling conditions are shown in Table 1. Similar thiosulfate concentrations and slightly higher gold extractions were observed when ore was milled
35 in SO₂ and ammonia mixtures.

Table 1: Gold extraction after 5 hours of milling

Gold extraction					
Conditions (all pH 10)	Residue	Calc. head grade	Gold extracted	% extracted	
	mg/Kg _{ore}	mg/Kg _{ore}	Mg/Kg _{ore}		
Experiment 1 (NH ₃ - O ₂ in solution)	35.01	67.9*	32.98	48.4	
Experiment 2 (NH ₃ - 1.2M SO ₂ in solution)	17.21	50.1	32.89	65.78	
Experiment 3 (NH ₃ - 0.5M SO ₂ in solution)	25.67	49.2	23.53	56.49	

*ore grade slightly higher than other samples used.

5 Example 2

This example demonstrates gold extraction following milling.

The ore used was a gold-copper sulfide concentrate that consisted mainly of pyrite (52%) quartz (15%), muscovite (11%), chlorite (2%), albite (5%), ankerite (12%), and traces of rutile (1.4%) and paragonite (0.5%). The ore concentrate contained 50 g/ton Au, 20 g/ton of Ag together with concentrations of Te (76 g/ton), Cu (2045 g/ton) and As (3370 g/ton). Mineralogical analysis showed the gold present in the ore existed as fine-grained inclusions in pyrite, as both coarse and fine-grained tellurides locked in and around pyrite grains, and as coarse and fine free milling gold. Approximately 65% of the gold could be extracted using the conventional cyanidation method, with high reagent consumption.

The ore was ground down to particle size having a P₈₀ of 10 micrometres. Prior to leaching, liquor from the slurry was tested for weak acid dissociable cyanide to demonstrate that no cyanide contaminants were present that would result in gold dissolution.

The generation of thiosulfate and leaching of

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gold from a slurry having 36% solids was carried out in a 5L stainless steel tank, with direct drive over-head stirrer maintained at a constant 600 rpm. Oxygen was sparged into the slurry at a flow rate of 100mL/min (corresponding to 2-4ppm dissolved oxygen) and the pH was maintained at 9.5 over the whole leaching period. Analysis for anion species in solution by HPLC confirmed the presence of up to 0.05M thiosulfate generated over a 24 hour period.

Evidence of gold and silver dissolution from thiosulfate generated from the leaching of ultra fine milled pyrite concentrate in oxygen at a pH of 9.5 is shown in Figure 2A.

The results show that about 50% of the gold was extracted after leaching for 4 hours (Figure 2a). Gold concentration in solution corresponds to free thiosulfate present in solution (~0.01mol/L, see Figure 2b). A maximum of 2mg/L copper leached from the ore was required to catalyse the gold leaching process. The calculated head, residue and % gold extraction for two experiments after 4 hours leaching are shown in Table 2.

Treatment of the sulfidic flotation concentrate without grinding resulted in no thiosulfate formation or detectable gold extraction.

Table 2: Gold extraction after 4 hours in duplicate leach experiments

Gold extraction (ore grade 50.1 mg/Kg)				
	Residue	Calc. head	Gold	% extracted
	mg/Kg _{ore}	mg/Kg _{ore}	mg/Kg _{ore}	
Experiment 1	24.1	49.9	25.9	51.8
Experiment 2	24.4	48.4	24.0	49.6

Example 3

This is a further example demonstrating gold extraction during milling. This example demonstrates that

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high gold extraction can be achieved with lower ammonia content.

The ore of Example 1 was ground in a batch 1.4 litre attritor mill for 5 hours under either mixtures of oxygen and 0.3M NaOH or 0.4M ammonia-oxygen and 0.15M NaOH in solution. Millings were carried out using 1-1.2 mm pebble media, a solids to water ratio of 1:2, a ball to sample weight ratio of 3:1, and an impeller tip speed of 6.4 m/s. In all experiments, the pH started at 12.0 and with time reach a pH of 6.0 after 3 hours. The dissolved oxygen levels were initially very low and increased over time (as reflected in the increase in gold extraction) and reduced. At the stage of best effect, the dissolved oxygen content was between 2 and 4 ppm (mg/L).

Figure 3 graphs the level of generation of thiosulfate and leaching of gold during the combined milling/extraction process in the presence of (a) oxygen and 0.3M NaOH or (b) 0.4M ammonia-oxygen and 0.15M NaOH in solution. (40% solids, 3:1 ball to charge ratio) and oxygen (0.5L/min). In figure 3a, the peak gold extraction peak correlates to a degree of extraction of about 35%, and in figure 3b the peak for gold extraction correlates to a percentage recovery of about 45%. These levels indicate that in a continuous process, high levels of gold recovery would be achievable.

Example 4

This example demonstrates gold extraction after milling using a three-stage leaching.

The ore of Example 1 was ground down to particle size having a P_{80} of 10 micrometres.

The generation of thiosulfate and leaching of gold from a slurry having 36% solids was carried out in a 5L stainless steel tank, with direct drive over-head stirrer maintained at a constant 600 rpm. Oxygen was sparged into the slurry at a flow rate of 100mL/min and the pH was maintained at 9.5 over the whole leaching period.

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Analysis for anion species in solution by HPLC confirmed the presence of up to 0.10M thiosulfate generated over a 3 hours. A three-stage oxidation leach was investigated. After three hours, the liquor was removed from the solid by
5 filter press and the residue washed with water. A second and then a third leach were carried out on the residue in the same way as in the first, for a further 3-hour period.

Table 3 shows the concentration of gold extracted and assays data from residue. This is also represented in
10 Figure 4. In the first stage, approximately 50% of the gold was leached after three hours. A further 35% and 9% of the remaining gold was removed in subsequent leaches. In all leaches there appeared to be sufficient free thiosulfate (0.025M) generated in situ available for complexing with
15 gold. Further leaching of the gold appears to have been prevented, and this is possibly related to the build up with time of trithionate present in all three leach solutions. Overall 70% of the gold has been leach from the ore without the addition of copper, ammonia or thiosulfate.
20 It is also noted that high gold extraction (25mg/kg of ore) is achieved after about 4 hours leaching.

This example demonstrates that in a commercial process, maximum recovery of gold can be achieved with multiple stage leaching, such as multiple stage
25 countercurrent leaching.

Table 3: Gold extracted and present in residue after three leaches

Experiment	Gold (mg/kg _{ore})
1 st leach	
Extracted	23.76
Residue	22.02
2 nd leach	
Extracted	7.74
Residue	15.6

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3 rd leach	
Extracted	1.4
Residue	14.00
Total calculated head grade	46.90
Recovered	70.14%

Example 5

This example demonstrates the effect of increase of alkalinity levels on the thiosulfate generation in different atmospheres.

The ore of Example 1 was ground in a batch 1.4 litre attritor mill up to 2 hours under different concentrations of caustic. Milling was carried out using 3-5 mm pebble media, a solids to water ratio of 1:2, a ball to sample weight ratio of 3:1, and an impeller tip speed of around 4.16 m/s. The pyrite concentrate was milled for up to 1.5 hours. Oxygen or air was injected into the airspace of the milling vessel at 1 L/min.

Figure 5 shows that significant concentrations of thiosulfate can be generated under the given conditions in the mill. Increased alkalinity results in an increased concentration of thiosulfate and the amount of thiosulfate generated is dependent upon the atmospheric conditions (i.e. oxygen or air). In addition, oxygen or air required to produce thiosulfate has to be controlled to prevent excessive oxidation of thiosulfate to trithionate, tetrathionate and sulfate, particularly when copper is present in the ore

It will be understood to persons skilled in the art of the invention that many modifications may be made without departing from the spirit and scope of the invention.

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Claims:

1. A process for leaching precious metals from an ore, the process comprising the steps of milling an ore
5 containing a precious metal to produce a milled ore, contacting the milled ore with water, an alkaline reagent, a suitable source of oxygen and a suitable source of sulfur in amounts sufficient to generate at least 0.02 mol/L thiosulfate in situ, and leaching the precious metal from
10 the milled ore into a thiosulfate-containing aqueous phase.
2. The process of claim 1, wherein at least 0.04 mol/L thiosulfate is generated in situ.
- 15 3. The process of claim 1 or claim 2, wherein the thiosulfate generated in situ constitutes from 30% to 100% of the thiosulfate present in the aqueous phase.
4. The process of claim 3, wherein the thiosulfate
20 generated in situ constitutes from 50% to 100% of the thiosulfate present in the aqueous phase.
5. A process for leaching precious metals from an ore, the process including the steps of milling an ore
25 containing a precious metal to produce a milled ore, contacting the milled ore with water, an alkaline reagent, a suitable source of oxygen and a suitable source of sulfur in amounts sufficient to generate thiosulfate in situ to form a thiosulfate-containing aqueous phase, and leaching
30 the precious metal from the milled ore into said aqueous phase, wherein the thiosulfate generated in situ constitutes at least 30% of the thiosulphate present in the aqueous phase.
- 35 6. The process of claim 5, wherein 0.02 mol/L of thiosulfate is generated in situ.

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7. The process of any one of claims 1 to 6, further comprising the step of recovering the precious metal from the aqueous phase.

5 8. The process of any one of the preceding claims, wherein the pH of the aqueous phase is between 7 and 11.

9. The process of any one of the preceding claims, wherein the alkaline reagent is selected ammonia, lime,
10 sodium hydroxide, potassium hydroxide, soda ash or potash.

10. The process of any one of the preceding claims, wherein the level of the source of oxygen present is sufficient to provide a dissolved oxygen content in the
15 aqueous phase of 1 mg/L, and up to 20 mg/L.

11. The process of claim 10, wherein the dissolved oxygen content is about 2 - 4 mg/L.

20 12. The process of any one of the preceding claims, wherein the ore is sulfide ore.

13. The process of claim 12, wherein the source of sulfur is the ore.

25 14. The process of any one of the preceding claims, wherein the source of sulfur includes elemental sulfur.

15. The process of any one of the preceding claims,
30 wherein the leaching is conducted in the presence of copper.

16. The process of any one of the preceding claims, wherein the leaching is conducted at a temperature of 70°C
35 or less.

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17. The process of claim 16, wherein the leaching is conducted at ambient temperature.

18. The process of any one of the preceding claims,
5 wherein the leaching is conducted at atmospheric pressure.

19. The process of any one of the preceding claims,
wherein the leaching is conducted over a period of between
1 and 48 hours.
10

20. The process of claim 19, wherein the leaching is conducted over a period of between 1 and 6 hours.

21. The process of any one of the preceding claims,
15 wherein the milling is conducted to produce a milled ore with a particle size having a P_{80} of 100 micrometres or less.

22. The process of claim 21, wherein the particle
20 size of the milled ore has a P_{80} of 50 micrometres or less.

23. A process for the recovery of precious metal from an ore, the process comprising the steps of milling an ore containing a precious metal to produce a milled ore,
25 contacting the milled ore with water, an alkaline reagent, a suitable source of oxygen and a suitable source of sulfur in amounts sufficient to generate at least 0.02 mol/L thiosulfate in situ, leaching the precious metal from the milled ore into a thiosulfate-containing aqueous phase, and
30 recovering the precious metal from the aqueous phase.

24. The process of claim 23, wherein the thiosulfate generated in situ constitutes from 30% to 100% of the thiosulfate present in the aqueous phase.
35

25. A precious metal recovered from the process of any one of the preceding claims.

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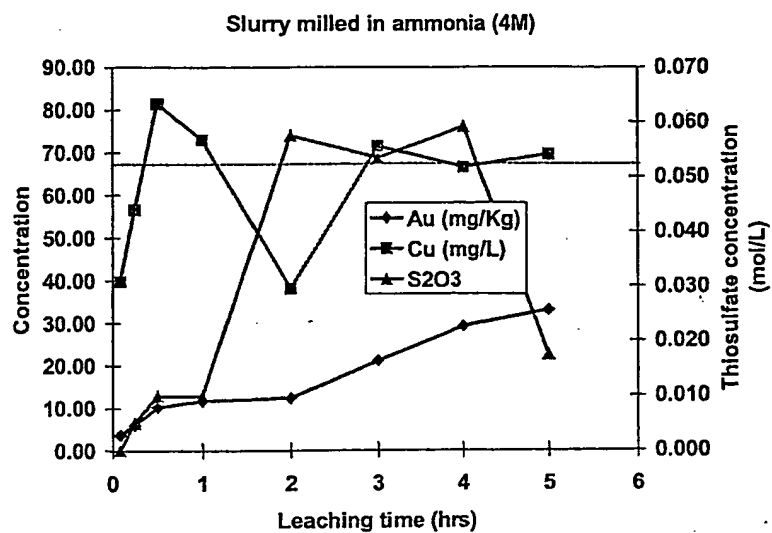


FIGURE 1

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Precious metals

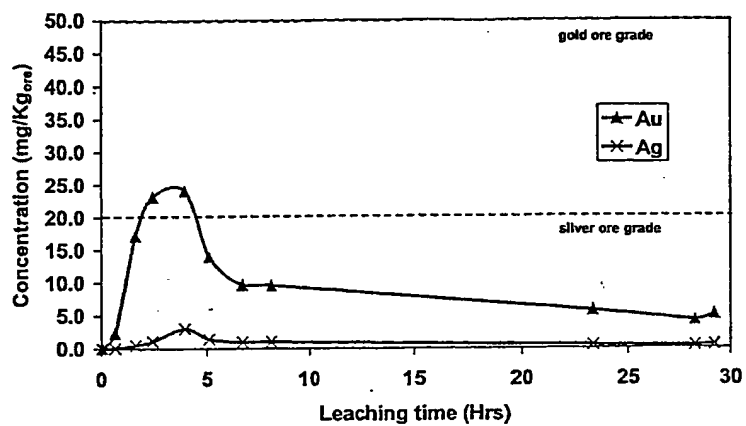


FIGURE 2a

Thiosulfate and polythionates

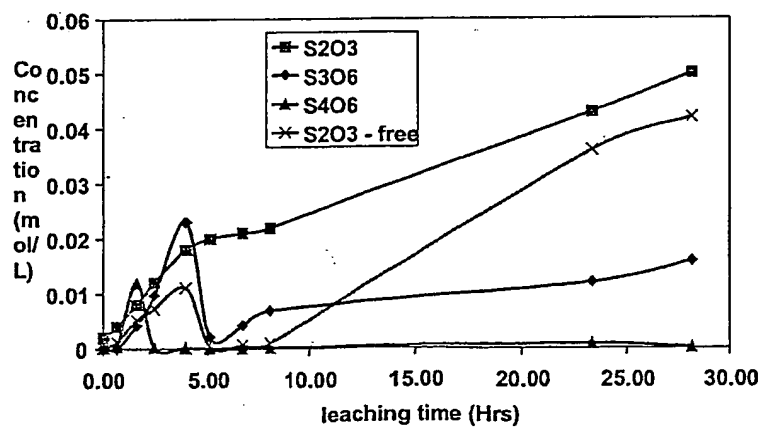


FIGURE 2b

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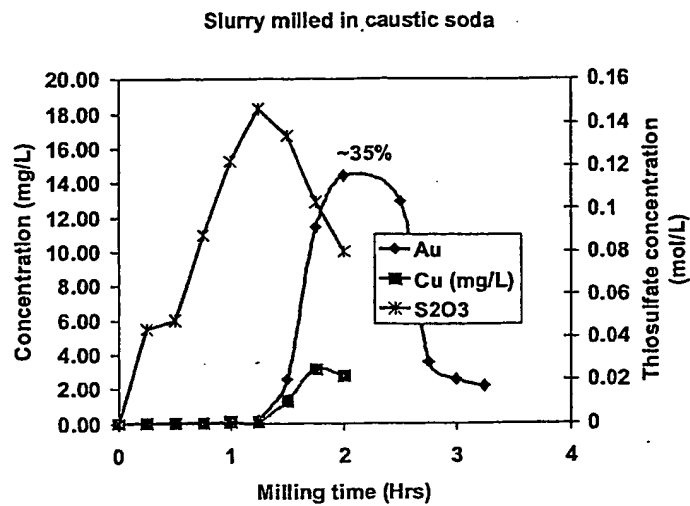


Figure 3A

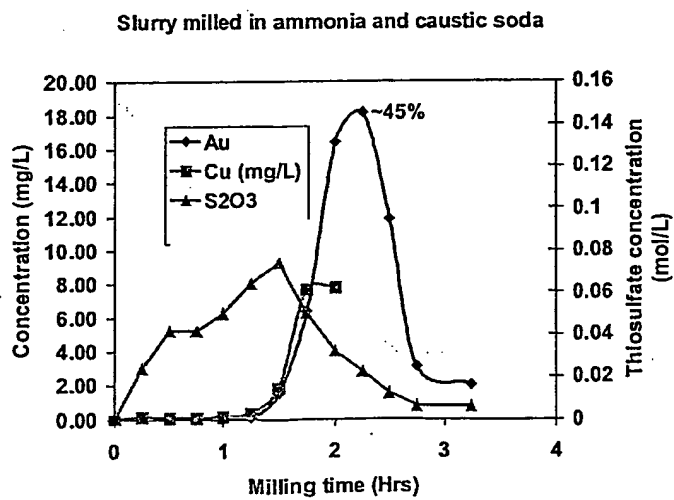


Figure 3B

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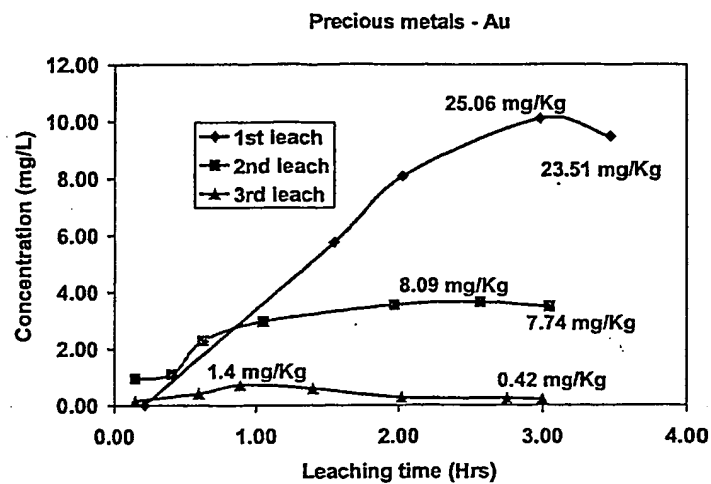


Figure 4

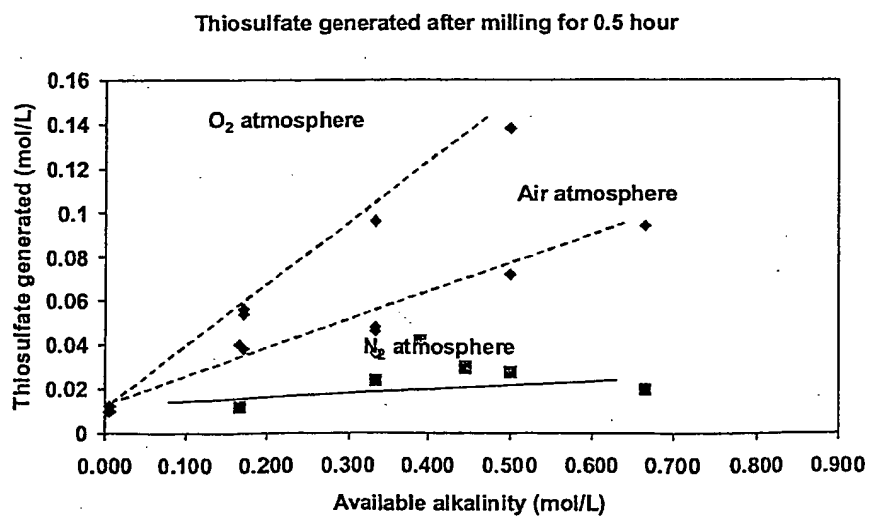


Figure 5

INTERNATIONAL SEARCH REPORT

International application No.
PCT/AU03/00855

A. CLASSIFICATION OF SUBJECT MATTER		
Int. Cl. ⁷ : C22B 3/04, 3/12, 3/14, 11/00		
According to International Patent Classification (IPC) or to both national classification and IPC		
B. FIELDS SEARCHED		
Minimum documentation searched (classification system followed by classification symbols)		
REFER ELECTRONIC DATA BASE CONSULTED		
Documentation searched other than minimum documentation to the extent that such documents are included in the fields searched		
Electronic data base consulted during the international search (name of data base and, where practicable, search terms used)		
DWPI, IPC: C22B 3/04, 3/12, 3/14, 11/00 and THIOSUL+		
C. DOCUMENTS CONSIDERED TO BE RELEVANT		
Category*	Citation of document, with indication, where appropriate, of the relevant passages	Relevant to claim No.
A	WO 01/88212 A (PLACER DOME TECHNICAL SERVICES LIMITED) 22 November 2001 whole document	1 - 25
A,P	AU 151812/01 A (THE UNIVERSITY OF BRITISH COLUMBIA) 24 October 2002 whole document	1 - 25
A	CA 2278044 A (PLACER DOME INC.) 19 January 2001 whole document	1 - 25
<input checked="" type="checkbox"/> Further documents are listed in the continuation of Box C <input checked="" type="checkbox"/> See patent family annex		
<p>* Special categories of cited documents:</p> <p>"A" document defining the general state of the art which is not considered to be of particular relevance</p> <p>"E" earlier application or patent but published on or after the international filing date</p> <p>"L" document which may throw doubts on priority claim(s) or which is cited to establish the publication date of another citation or other special reason (as specified)</p> <p>"O" document referring to an oral disclosure, use, exhibition or other means</p> <p>"P" document published prior to the international filing date but later than the priority date claimed</p> <p>"T" later document published after the international filing date or priority date and not in conflict with the application but cited to understand the principle or theory underlying the invention</p> <p>"X" document of particular relevance; the claimed invention cannot be considered novel or cannot be considered to involve an inventive step when the document is taken alone</p> <p>"Y" document of particular relevance; the claimed invention cannot be considered to involve an inventive step when the document is combined with one or more other such documents, such combination being obvious to a person skilled in the art</p> <p>"&" document member of the same patent family</p>		
Date of the actual completion of the international search 19 August 2003		Date of mailing of the international search report 28 AUG 2003
Name and mailing address of the ISA/AU AUSTRALIAN PATENT OFFICE PO BOX 200, WODEN ACT 2606, AUSTRALIA E-mail address: pct@ipaustralia.gov.au Facsimile No. (02) 6285 3929		Authorized officer B. PREMARATNE Telephone No : (02) 6283 2407

INTERNATIONAL SEARCH REPORT

International application No.

PCT/AU03/00855

C (Continuation).

DOCUMENTS CONSIDERED TO BE RELEVANT

Category*	Citation of document, with indication, where appropriate, of the relevant passages	Relevant to claim No.
A	US 4445935 A (POSEY et al.) 1 May 1984 whole document	1 - 25
A	US 5536297 A (MARCHBANK et al.) 16 July 1996 whole document	1 - 25

INTERNATIONAL SEARCH REPORT

Information on patent family members

International application No.

PCT/AU03/00855

This Annex lists the known "A" publication level patent family members relating to the patent documents cited in the above-mentioned international search report. The Australian Patent Office is in no way liable for these particulars which are merely given for the purpose of information.

Patent Document Cited in Search Report		Patent Family Member			
WO	200188212	AU	200174393	CA	2409378
		US	2002092377		
AU	51812/01	AU	200151812	US	2003051581
CA	2278044	NONE			
US	4445935	CA	1214334	DE	3345022
		JP	59113133		
US	5536297	CA	2169140	US	5785736
				CA	2209559
END OF ANNEX					